Energy, Mines and Resources Canada
Énergie, Mines et Ressources Canada

CANMET
Canada Centre for Mineral and Energy Technology
Centre canadien de la technologie des minéraux et de l'énergie

PRELIMINARY BENCH-SCALE METALLURGICAL INVESTIGATION KIDD CREEK "C" ORE

A.I. Stemerowicz and G.W. Leigh
Mineral Processing Laboratory

December 1988

Project: 30.13.06
Beneficiation of Complex Ores

MICROMEDIA

58pp

MINERAL SCIENCES LABORATORIES
DIVISION REPORT MSL 88-151 (IR)

PRELIMINARY BENCH-SCALE METALLURGICAL INVESTIGATION KIDD CREEK "C" ORE

by

A.I. Stemerowicz* and G.W. Leigh**

SUMMARY

A preliminary investigation consisting of 12 bench-scale tests was carried out on a sample of Kidd Creek "C" ore assaying 0.49% Cu, 1.70% Pb, 9.20% Zn and 26.7.3 g/t Ag. The main problem encountered was the difficulty in achieving a satisfactory degree of selectivity between chalcopyrite and pyrite and galena and pyrite.

Three methods were tried for the production of copper and lead concentrates: (1) sequential Cu-Pb-Zn selective flotation from the ore which is currently being employed by Kidd Creek (2) copper-lead separation of a bulk copper-lead concentrate and (3) copper-lead-zinc separation of a bulk copper-lead-zinc concentrate. In addition an alternative scheme was also tried in which the ground ore was first subjected to a 21-hour period of cyanidation at a cyanide solution strength of 1 g/L followed by the flotation of lead and zinc concentrates from the cyanide residue. At a grind of 89.3% minus 44 µm this scheme gave a lead concentrate grade of 47.4% lead with 62.8% recovery and a zinc concentrate grade of 56.2% zinc with 85.3% recovery. These results are far superior to those obtained using the other, conventional selective flotation methods. Total silver recovery in the pregnant solution and lead and zinc concentrates was 80.1% compared to the 82.7% silver recovered in the copper, lead and zinc concentrates at the Kidd Creek plant. In the one test in which this scheme was employed, practically all of the copper was rejected to the final tailing instead of floating with the zinc as was anticipated. However, it is believed that the chalcopyrite can be reactivated by intense aeration of the lead rougher tailing and floated off prior to zinc flotation. Because some of the silver

---

* Research Scientist and **Technician, Mineral Processing Laboratory
CANMET, 555 Booth Street, Ottawa, Ontario, K1A 0G1, Canada

in the ore is associated with chalcopyrite the recovery of a copper concentrate will be accompanied by an increase in overall silver recovery. Continuation of the investigation along these lines is recommended along with quantitative mineralogical study of the ore and test products.

Keywords: Mineral processing, Concentration, Complex sulphide ores
Selective sulphide flotation, Kidd Creek

ANALYSE METALLURGIQUE DE LABORATOIRE PRELIMINAIRE MINERAI "C" DE KIDD CREEK

par

A.I. Stemerowicz* et G.W. Leigh**

RÉSUMÉ

Une analyse préliminaire comportant 12 essais en laboratoire a été effectuée sur un échantillon de minerai "C" de Kidd Creek titrant 0,49% Cu, 1,7% Pb, 9,20% Zn et 26,7,3 g/t Ag. Le principal problème qui s'est posé a été la difficulté de réaliser un degré satisfaisant de sélectivité entre la chalcopyrite et la pyrite, et la galne et la pyrite.

Trois méthodes ont été essayées pour produire des concentrés de cuivre et de plomb: (1) flotation sélective séquentielle Cu-Pb-Zn du minerai qui est actuellement utilisé par Kidd Creek, (2) séparation cuivre-plomb d'un concentré mixte cuivre-plomb et (3) séparation cuivre-plomb-zinc d'un concentré mixte cuivre-plomb-zinc. On a aussi essayé une méthode de remplacement dans laquelle le minerai broyé était d'abord soumis à une période de cyanuration de 21 heures dans une solution titrant 1 g/L de cyanure, suivie de la flottation des concentrés de plomb et de zinc provenant des résidus cyanurés. A un niveau de broyage de 89,3% à moins 44 µm, cette méthode a produit un concentré titrant 47,4% de plomb avec une récupération de 62,8% et un concentré titrant 56,2% de zinc avec une récupération de 85,3%. Ces résultats sont de beaucoup supérieurs à ceux obtenus au moyen des autres méthodes classiques de flottation sélective. La récupération d'argent total dans la solution enrichie et les concentrés de plomb et de zinc a été de 80,1% comparativement à 82,7% d'argent récupéré dans les concentrés de cuivre, de plomb et de zinc à l'usine de Kidd Creek. Dans le seul essai où cette méthode a été employée, presque tout le cuivre a été rejeté dans le résidu final au lieu de flotter avec le zinc comme prévu. On estime toutefois que la chalcopyrite peut être réactivée par une aération intense du résidu de dégrossissage du minerai de plomb et éliminée

par flottation avant la flottation du zinc. Comme une part de l'argent contenu dans le minerai est associée à la chalcopyrite, la récupération d'un concentré de cuivre sera accompagnée par une augmentation de la récupération totale d'argent. On recommande de poursuivre les recherches dans cette voie ainsi que l'étude minéralogique du minerai et des produits d'essai.

Mot clés: traitement des minéraux, concentration, minerais sulfurés complexes, flotation sélective des sulfures, Kidd Creek

CONTENTS

Page

SUMMARY.....i
RÉSUMÉ.....iii
INTRODUCTION.....1
OUTLINE OF INVESTIGATION.....2
  Ore Sample.....2
  Crushing and Sampling.....2
  Grinding.....2
  Flotation.....3
  Test Data.....6
DISCUSSION AND EVALUATION OF RESULTS.....7
  Best Results.....7
  Cyanidation of Reground Pyrite Concentrate Followed
    by Selective Flotation from the Residue (Test C-10).....7
  Sequential Cu-Pb-Zn Selective Flotation.....9
  Flotation of a Copper-Lead Concentrate Followed
    by copper-lead separation.....10
  Cu-Pb-Zn Bulk Flotation Followed by Regrinding Cleaning
    and Separation to Produce Separate
    Copper, Lead and Zinc Concentrates.....16
CONCLUSIONS.....17
REFERENCES.....18
APPENDIX.....19

TABLES

No.

1. Typical Kidd Creek plant results for sequential Cu-Pb-Zn selective flotation of "C" ore.....1
2. Analysis of head sample.....2
3. Particle size distribution for various grinding times.....3
4. Results for cyanidation followed by lead and zinc flotation (Test C-9).....7

TABLES (cont'd)

No. Page

5. Results for cyanidation of reground pyrite concentrate followed by selective flotation from cyanide residue (Test C-10).....8
6. Separation efficiency for zinc flotation from pyrite tailing (Test C-10).....9
7. Results for sequential Cu-Pb-Zn selective flotation using F2B as copper collector.....10
8. Comparison of copper-lead rougher flotation results.....11
9. Typical copper-lead cleaning results (Test C-11).....11
10. Comparison of copper-lead separation results for copper rougher flotation using various techniques.....13
11. Copper-lead separation results for Test C-11.....14
12. Comparison of copper and lead concentrate grades and recoveries - Test C-11 versus Kidd Creek plant results.....14
13. Results for flotation of a secondary zinc concentrate from the copper-lead separation tailing (Test C-6).....15
14. Results for the production of separate concentrates from a bulk concentrate.....16

INTRODUCTION

This investigation is the initial project element of the Mineral Sciences Laboratories project entitled "Beneficiation of Complex and Refractory Ores".

Currently the Kidd Creek Cu-Pb-Zn-Ag ore ("C" ore) is concentrated by utilizing sequential copper-lead-zinc selective flotation. Typical results are given in Table 1 (1).

Table 1. Typical Kidd Creek plant results for sequential Cu-Pb-Zn selective flotation of "C" ore

Product	Wt %	Analysis				Distribution %			
		Cu %	Pb %	Zn %	Ag g/t	Cu	Pb	Zn	Ag
Cu conc	3.17	18.12	9.62	4.30	5412.0	63.5	26.2	1.9	58.6
Pb conc	3.36	3.84	15.79	7.46	1293.0	14.2	45.5	3.5	14.8
Zn conc	11.23	0.71	1.10	50.49	243.0	8.8	10.6	78.8	9.3
Zn ro tail	82.24	0.15	0.25	1.38	61.80	13.5	17.7	15.8	17.3
Feed (calcd)	100.00	0.91	1.17	7.19	293.0	100.0	100.0	100.0	100.0

These results were obtained at a primary grind of 73% minus 45 µm and with zinc middlings (zinc scavenger concentrate + zinc cleaner tailings) reground prior to recycling. As can be seen there is much scope for improving concentrate grades and recoveries. It was for this reason that this ore was chosen as the first to be investigated.

The specific objectives in this first series of exploratory concentration tests were to determine (1) the effect of very fine grinding and regrinding (2) the response of the ore to depressants such as sulphur dioxide and cyanide and (3) to try alternatives to sequential copper-lead zinc flotation currently employed in the Kidd Creek concentrator.

OUTLINE OF INVESTIGATION

Ore Sample

A 42 kg sample (dry weight) of Kidd Creek "C" ore was received on February 18, 1988. It consisted of damp rod mill feed containing on estimated 1 to 2% moisture.

Crushing and Sampling

After removing the minus 10 mesh fines (1.68 mm) the wet ore sample was crushed in stages to minus 10 mesh. Both ore fractions were then dried at -30°C in a freeze drier under high vacuum and in the presence of dessicant. The two dry ore fractions were blended and split into 20 portions using a carrousel-type divider. Each of the portions was then adjusted to exactly 2000 grams by removing the excess material. This excess material amounting to 1985 grams was designated as the head sample. To prevent oxidation the 2000-g lots were stored in the freeze drier at high vacuum and with dessicant. Analysis of the head sample is given in Table 2.

Table 2. Analysis of head sample

Cu	0.49%
Pb	1.70%
Zn	9.20%
Ag	267.3 g/t
Total Fe	19.00%
S	20.69%
Insolubles	35.6%

Grinding

The 2000-g minus 10 mesh test charges were ground at 65% solids in a 12-in diameter (305 mm) Abbé ceramic ball mill with a charge of 19 Kg of 25 mm diameter steel balls in the first grinding stage and 5.9 Kg of minus 16 mm ceramic balls in subsequent stages. Regrinding of test products was carried out at 50% solids in an 8-in diameter (203 mm) Abbé ceramic ball mill with half the mill volume occupied by a charge of minus 16 mm ceramic balls. Particle size distributions obtained for the various grinding times employed are given in Table 3.

Table 3. Particle size distribution for various grinding times

Tyler mesh size	µm	50 min (1) 1 stage grind		60 min (2) 1 stage grind		90 min (3) 1 stage grind		3x30 min (4) 3 stage grind	
		% ret	% pass	% ret	% pass	% ret	% pass	% ret	% pass
+100	149	0.1	99.9	-	-	-	-	-	-
+200	74	2.0	97.9	0.4	99.6	0.1	99.9	0.1	99.9
+270	53	3.2	94.7	-	-	0.2	99.7	0.1	99.8
+325	44	5.4	89.3	2.9	96.7	1.0	98.7	0.3	99.5
+400	37	8.2	81.1	5.2	91.5	2.7	96.0	1.5	98.0
+500	25	11.3	69.8	10.6	80.8	6.9	89.1	6.3	91.7
-500		69.8		80.8		89.1		91.7	
Total		100.0		100.0		100.0		100.0	

Notes: (1) Tests C-8 to C-12
(2) Tests C-2 and C-3
(3) Tests C-4 to C-6
(4) Test C-1 - Size distribution given is for zinc rougher tailing

Flotation

Sequential Cu-Pb-Zn Selective Flotation

The starting point of the investigation was to carry out a few sequential Cu-Pb-Zn selective flotation tests to compare results with those obtained at Kidd Creek. Three tests were carried out as follows:

Test C-1 - In this test a very fine 3-stage grind of about 92% minus 25 µm was employed (size distribution of zinc rougher tailing). The first two grinding stages were on test feed while the third was a regrind of copper rougher tailing prior to lead flotation. Sulphur dioxide at a pH 5.4 was employed as the galena and sphalerite depressant during the initial copper rougher flotation step. After floating off a copper rougher concentrate with a combination of isobutyl xanthate and Aerofloat 238 as collectors the

copper rougher tailing was reground with soda ash and cyanide added to reactivate the galena. Lead flotation was carried out at a pH of 7.9 with a combination of isobutyl xanthate and Aerofloat 31 as collectors. The lead rougher tailing was conditioned with 100 g/t copper sulphate and lime for a pH of 11 and a zinc rougher concentrate was floated off using mostly sodium aerofloat as collector augmented by a small amount of isobutyl xanthate. This procedure differs from Kidd Creek plant practice in 2 important respects:

(1) At Kidd Creek lime is added in addition to sulphur dioxide to give a copper rougher flotation pH of 8.5.

(2) Lead rougher flotation is conducted at a pH of 8.2 without a prior galena reactivation step.

Test C-2 - This was a repeat of Test C-1 but at a coarser grind of 81% minus 25 µm and without a galena reactivation step prior to lead flotation.

Test C-4 - The purpose of this test, which was carried out at a very fine single-stage grind of 89% minus 25 µm, was to try Dow Chemical Co. development collector F2B as a copper collector in conjunction with Aerofloat 208. Alternative collectors were also used for lead and zinc flotation, viz., Aerofloat 241 in place of Aerofloat 31 and isopropyl xanthate instead of isobutyl xanthate.

Flotation of a Copper-Lead Concentrate Followed by Copper-Lead Separation

This is the preferred and generally the most efficient method of producing separate copper and lead concentrates from most Cu-Pb-Zn sulphide ores.

Five flotation tests were carried out utilizing this method. In Tests C-3, 11 and 12 conventional techniques were employed for copper-lead flotation at a primary grind of 81% minus 25 µm in Test C-3 and 70% minus 25 µm in Tests C-11 and 12 while in Tests C-5 and 6 the Dow Chemical Co. collector F2B was again tried as a copper collector in conjunction with isopropyl xanthate and Aerophine 3418A which are also effective lead collectors. In Tests C-3, 11 and 12 the copper-lead rougher concentrate was reground and cleaned prior to the copper-lead separation step while in C-5

and 6 copper-lead separation was carried out directly on the reground copper-lead rougher concentrate. In all tests the object was to depress galena using either sodium dichromate or sulphur dioxide and then float off the unaffected chalcopyrite. Separation techniques varied from test to test as follows:

Test C-3 - The pulp was heated to 70°C and conditioned for 5 minutes with 200 g/t sodium dichromate and 75 g/t sulphur dioxide to lower the pH to 8.0.

Test C-11 - The collector was desorbed from the sulphide surfaces prior to the separation step by conditioning the copper-lead cleaner concentrate for 10 min with 2000 g/t NaOH at a pulp temperature of 95°C. After the desorption step the pulp was filtered and the cake washed free of solution. Separation was carried out at room temperature using 400 g/t dichromate augmented by 125 g/t CMC as galena depressants.

Test C-12 - Repeat of Test C-11 except that acetone was tried as collector desorbant in place of NaOH + heat.

Test C-5 - Sodium dichromate was added to the copper-lead rougher concentrate regrinding step. After copper flotation the separation tailing was subjected to zinc flotation to recover a secondary zinc rougher concentrate.

Test C-6 - Similar to Test C-5 except that sulphur dioxide at a pH of 5.2 was employed in place of dichromate as the lead depressant and separation was carried out at a pulp temperature of 80°C.

Cu-Pb-Zn Bulk Flotation Followed by Regrinding, Cleaning and Separation to Produce Copper, Lead and Zinc Concentrates

This technique was tried in Test C-8. To recover all of the valuable sulphides in a single bulk concentrate the ground ore was subjected to zinc flotation. The bulk rougher concentrate was then reground and cleaned twice with lime added for a high pH (11+) to depress galena and pyrite. The bulk cleaner concentrate was subjected to copper-zinc separation using sulphur dioxide at a pH of 4.8 and pulp temperature of 86°C

as the sphalerite depressant. This was followed by the flotation of a lead concentrate from the combined bulk cleaner tailings after the galena had been reactivated in a regrinding step with soda ash and cyanide.

Cyanidation of the Ore Followed by Lead and Zinc Flotation from the Cyanide Residue

This method was tried in Test C-9. The object was to take advantage of the severe depressing effect of NaCN on pyrite when it is subjected to a high cyanide concentration for a prolonged period. The ground ore (70% minus 25 µm) was cyanided for 21 hours at a cyanide strength of 1 g/L. The cyanide residue which had been filtered and washed free of solution was then subjected to lead flotation followed by zinc flotation. It was assumed that most of the chalcopyrite would be reactivated by copper sulphate and be recovered in the zinc concentrate from which a copper concentrate could be produced by employing the sulphur dioxide + hot pulp method for copper-zinc separation. The silver recovered by cyanidation would be precipitated from the pregnant solution as a high-grade product.

Cyanidation of a Reground Pyrite Concentrate Floated from the Ore Followed by Copper-Lead and Zinc Flotation from the Cyanide Residue

The object of this technique which was tested in Test C-10 was to maximize recoveries by floating off and regrinding a pyrite concentrate to liberate the valuable minerals closely associated with pyrite. In Test C-9 these composite particles would most likely have been rejected to the tailings. Pyrite flotation was carried out at a natural pH of 8.3 with stage additions of a total of 133 g/t collector. Under these conditions high recoveries of copper and lead in the pyrite concentrate could be expected. It was anticipated that only a portion of the unactivated sphalerite would float with pyrite - the balance was recovered by floating a zinc concentrate from the pyrite tailing. This product was referred to as the "primary zinc concentrate" to distinguish it from the "secondary zinc concentrate" floated from the cyanide residue.

Test Data

Detailed flotation test data and metallurgical results for all the tests described are appended. Also included, because of their complexity, are flowsheets for tests C-8 and C-10.

DISCUSSION AND EVALUATION OF RESULTS

Best Results

The best results for lead and zinc flotation were obtained in Test C-9 in which the ore was ground to 89.3% minus 44 µm, cyanided for 21 hours and the cyanide residue subjected to lead and zinc flotation. Results are summarized in Table 4.

Table 4. Results for cyanidation followed by lead and zinc flotation (Test C-9)

Product	Analysis				Distribution %			
	Cu%	Pb %	Zn %	Ag g/t	Cu	Pb	Zn	Ag
Pregnant soln	-	-	-	-	-	-	-	56.1
Lead conc	0.48	47.40	6.50	1716.0	2.2	62.8	1.9	17.1
Zinc conc	0.34	0.86	56.20	130.6	8.4	6.0	85.3	6.9
Lead + Zinc cl tail	-	-	-	-	17.8	12.7	7.7	7.9
Zinc ro tail	0.54	0.49	0.62	42.3	71.6	18.5	5.1	12.0

As can be seen the beneficial effect of the long contact of the ground ore with cyanide on subsequent lead and zinc flotation was confirmed. Galena-pyrite and sphalerite-pyrite selectivity was much superior to that obtained using the other conventional selective flotation techniques. With regard to silver recovery there was no advantage in using this scheme. The total silver recovered in the pregnant solution and lead concentrate was about the same as the sum of the silver recoveries in the Kidd Creek plant copper and lead concentrates. It was not possible to produce a copper concentrate by copper-zinc separation of the zinc concentrate because most of the copper was rejected to the tailing instead of floating with the zinc as was expected.

Cyanidation of Reground Pyrite Concentrate Followed by Selective Flotation from the Residue (Test C-10)

As was stated previously the objective was to maximize recoveries by liberating valuable minerals closely associated with pyrite. In Test C-9 this type of middling particle would most likely have been rejected to the zinc rougher tailing. Results are given in Table 5.

Table 5. Results for cyanidation of reground pyrite concentrate followed by selective flotation from cyanide residue (Test C-10)

Product	Wt %	Analysis %				Distribution %			
		Cu %	Pb %	Zn %	Ag g/t	Cu %	Pb %	Zn %	Ag g/t
Preg soln	-	-	-	-	-	1.4	-	0.1	35.5
Cu-Pb ro conc	16.63	2.27	8.00	8.82	545.7	63.8	70.6	15.9	36.7
Sec Zn ro conc	7.83	1.15	2.74	34.11	328.2	15.1	11.4	29.0	10.5
Sec Zn ro tail	17.58	0.35	1.10	0.66	74.7	10.4	10.3	1.3	5.3
Prim Zn ro conc	11.49	0.26	0.47	41.87	106.2	4.9	2.8	52.2	4.9
Prim Zn ro tail	46.47	0.056	0.20	0.29	37.9	4.4	4.9	1.5	7.1
Feed (calcd)	100.00	0.60	1.89	9.21	247.2	100.0	100.0	100.0	100.0
Pyrite conc	42.04	1.29	4.14	10.13	51.70	90.7	92.3	46.3	88.0
Prim Zn cl conc	8.15	0.22	0.33	57.30	101.5	3.0	1.4	50.7	3.3
Sec Zn cl conc	5.64	0.98	2.80	46.00	387.9	8.8	8.4	28.2	8.9
Zn conc comb	13.79	0.51	1.34	52.68	218.6	11.8	9.8	78.9	12.2
Zn ro tail comb	64.05	0.14	0.45	0.39	48.0	14.8	15.2	2.8	12.4

In contrast to the previous test on ore cyanide residue, selectivity against pyrite and sphalerite during copper-lead flotation from the pyrite cyanide residue was unsatisfactory. This was anticipated from the appearance of the copper-lead rougher flotation froth which consisted of a mass of very fine mineralized bubbles in the form of voluminous and watery foam. The authors have encountered this condition previously in investigations of other sulphide ores. It is generally a result of the interaction between cyanide and dithiophosphate type collectors which were used in this test in conjunction

with xanthate for pyrite flotation. It was thought that the foamy, non-selective type of froth could be avoided by removing and discarding solution prior to and after the regrinding step and by drying the pyrite concentrate prior to cyanidation. Perhaps the very finely-ground nature of the pyrite cyanide residue had a synergistic effect in producing this type of foamy froth.

It should be pointed out that in order to promote the flotation of chalcopyrite from the pyrite cyanide residue, the pulp was aerated for 30 minutes prior to flotation. In Test C-9 the aeration step was intentionally omitted because the aim was to produce a low-copper lead concentrate.

Worthy of note is the unusually high separation efficiency obtained for the flotation of zinc from the pyrite tailing as illustrated in Table 6. In this table the calculation of sphalerite content is based on an average iron content of 6.0% as determined by the Process Mineralogy Section. Thus pure sphalerite is assumed to have the following composition: Zn 61%, Fe 6%, S 33%.

Table 6. Separation efficiency for zinc flotation from pyrite tailing (Test C-10)

Product	Analysis %			Distribution %			Sep * eff %
	Zn %	Sphal %	Gangue %	Zn %	Sphal %	Gangue %	
Zn cl conc 2	57.30	93.9	6.1	94.4	94.4	1.0	93.4
Zn cl conc 1	54.64	89.6	10.4	95.6	95.6	1.8	93.8
Zn ro conc	41.87	68.6	31.4	97.3	97.3	7.2	90.1

* Recovery of sphalerite - recovery of gangue

When the primary zinc concentrate floated from the pyrite tailing is combined with the secondary zinc concentrate floated from the pyrite cyanide residue the resultant concentrate has a calculated grade of 52.68% Zn with 78.9% recovery.

Sequential Cu-Pb-Zn Selective Flotation

Generally for most sulphide ores, sulphur dioxide is an effective depressant for galena during copper flotation, especially when selective copper collectors are employed. This ore appears to be an exception. In

Test C-1 in which the first attempt was made at sequential Cu-Pb-Zn selective flotation, sulphur dioxide was employed at a pH of 5.3 and was found to be ineffective as a galena depressant. The copper rougher concentrate assayed 4.06% Cu, 10.16% Pb and 7.77% Zn with 68.4% of the lead recovered in the concentrate. Selectivity against pyrite was also very poor but this is a problem common to all fine-grained complex sulphide ores.

In Test C-4 sequential selective flotation was tried using Dow Chemical Co. developmental collector F2B which is recommended as a good chalcopyrite collector with excellent selectivity over pyrite. To ensure recovery of the silver minerals in the copper concentrate Aerofloat 208 was used in conjunction with F2B. Results are summarized in Table 7.

Table 7. Results for sequential Cu-Pb-Zn selective flotation using F2B as copper collector

Product	Analysis				Distribution %			
	Cu %	Pb %	Zn %	Ag g/t	Cu	Pb	Zn	Ag
Cu ro conc comb	1.55	3.20	8.91	1994.5	19.2	12.0	7.2	53.1
Pb ro conc	2.12	23.31	7.10	1111.5	19.3	63.5	4.2	21.6
Zn ro + scan conc	1.53	1.11	37.46	182.7	53.4	11.6	84.3	13.7
Zn ro tail	0.07	0.37	0.58	46.6	8.1	12.9	4.3	11.6
Feed (calcd)	0.58	1.93	8.96	270.0	100.0	100.0	100.0	100.0

As can be seen the collector F2B was ineffective as a chalcopyrite collector and did not exhibit any selectivity against pyrite. Most of the copper was recovered in the zinc rougher and scavenger concentrates.

Flotation of a Copper-lead Concentrate Followed by Copper-Lead Separation

Copper-Lead Rougher Flotation

Table 8 compares copper-lead rougher flotation results obtained using conventional techniques (Tests C-3, 11 and 12) with those obtained using the Dow developmental collector F2B in conjunction with isopropyl xanthate and Aerophine 3418A at a pH of 9.5 (Tests C-5 and C-6). In these two tests no depressants other than lime were added to the grind.

As was found previously for Cu-Pb-Zn sequential flotation, F2B was ineffective as a copper collector. Copper recoveries were about 10% lower even though the total amount of collector employed (F2B + isopropyl xanthate + Aerophine 3418A) was higher (39 g/t) than the Aerofloat 241 + Aerophine 3418A combination (35 g/t) employed in Tests C-3, 11 and 12.

Table 8. Comparison of copper-lead rougher flotation results

Test No.	Depressants and Collectors employed	Analysis				Distribution %			
		Cu %	Pb %	Zn %	Ag g/t	Cu	Pb	Zn	Ag
C-3	NaCN, $ZnSO_4$, pH 8.5* AF 241, Aero 3418A	3.05	9.38	8.84	1210.4	88.6	87.4	17.3	80.7
C-5	Lime, pH 9.5 F2B, 343, Aero 3418A	3.90	7.98	8.80	1572.8	79.7	63.3	12.3	73.9
C-6	Lime, pH 9.5 F2B, 343, Aero 3418A	2.10	7.97	9.34	982.1	82.9	79.1	20.5	77.8
C-11	As in Test C-3	3.25	10.31	8.97	1273.9	91.5	87.3	15.5	81.3
C-12	As in Test C-3	3.04	9.54	9.08	1187.9	92.9	88.9	18.2	82.3

* Soda ash added to grind

Note the low grade nature of the copper-lead rougher concentrate due primarily to poor selectivity against pyrite. For example in Test C-11 chalcopyrite, galena and sphalerite only account for 36% of the content of the copper-lead rougher concentrate. The balance of 64% is made up mostly of pyrite.

Copper-Lead Cleaning

In Tests C-3, 11 and 12 the copper-lead rougher concentrate was reground and cleaned twice prior to the copper-lead separation step. To ensure depression of pyrite and sphalerite 50 g/t NaCN was added to the regrind mill along with 250 g/t soda ash for alkalinity control. Typical copper-lead cleaning results are given in Table 9.

Most of the upgrading occurred in the first cleaning stage with 67.5% of the zinc and 69.4% of the pyrite rejected to the first cleaner tailing. In the second cleaning stage about 90% of the weight of the second

cleaner feed was recovered in the concentrate with the result that only a minimal amount of upgrading was achieved. This indicates the need for a further addition of cyanide to depress pyrite in the second cleaning stage.

Table 9. Typical copper-lead cleaning results (Test C-11)

Product	Wt %	Analysis			Distribution %						
		Cu	Pb	Zn	Ag	Pyrite	Cu	Pb	Zn	Ag	Pyrite
Cu-Pb cl conc 2	39.53	7.29	24.32	6.01	2915.0	41.0	88.8	93.3	26.5	90.5	25.3
Cu-Pb cl tail 2	4.44	0.78	1.60	12.20	273.4	75.9	1.0	0.7	6.0	0.9	5.3
Cu-Pb cl tail 1	56.03	0.59	1.10	10.80	195.3	79.3	10.2	6.0	67.5	8.6	69.4
Feed	100.0	3.25	10.31	8.97	1273.9	64.0	100.0	100.0	100.0	100.0	100.0
Cu-Pb cl conc 1	43.97	6.64	22.05	6.64	2648.4	44.5	89.8	94.0	32.5	91.4	30.6

Copper-Lead Separation

Table 10 compares results for copper-lead separation using various techniques. The comparison is made only for the initial flotation of the copper rougher concentrate from the reground copper-lead rougher concentrate in Tests C-5 and 6 and the copper-lead cleaner concentrate in Tests C-13, 11 and 12.

The best copper-lead separation efficiency that could be achieved was an unsatisfactory 56.4% in Test C-6 on the reground copper-rougher concentrate. In Test C-6 sulphur dioxide added to the regrinding step was employed as the galena depressant in conjunction with a hot pulp. Note that sulphur dioxide is a more effective galena depressant when added to regrinding than when added to a pulp conditioning step as was the case for sequential copper-lead-zinc flotation (Tests C-1 and 2). Sulphur dioxide had two other important effects when compared to Test C-5 a parallel test in which sodium dichromate was employed as the galena depressant:
(1) it promoted the flotation of pyrite which resulted in a low grade copper rougher concentrate and (2) it had a depressing effect on the silver minerals.

Table 10. Comparison of copper-lead separation results for copper rougher flotation using various techniques

Test No.	Reagents added	g/t	Desorpn step	Hot pulp	Product	Wt %	Analysis %				Distribution %				SE% Cu/Pb
							Cu %	Pb %	Zn %	Ag g/t	Cu	Pb	Zn	Ag	
C-3	$K_2Cr_2O_7$, SO_2	200 75 pH 8.0	No	Yes 70°C	Cu ro conc	64.52	10.75	23.74	6.38	3736	93.5	65.6	65.8	85.9	27.9
					Sepn tail	35.48	1.36	22.66	6.04	1116	6.5	34.4	34.2	14.1	
					Feed (Cu-Pb cl conc)	100.00	7.41	23.36	6.26	2806	100.0	100.0	100.0	100.0	
C-5	$K_2Cr_2O_7$,* AF 208 * to RG mill	400 18 pH 8.8	No	No	Cu ro conc-1	19.36	11.98	6.79	8.40	5287	59.4	16.5	18.5	65.1	42.9
					Cu ro conc-2	8.40	4.91	10.61	12.44	2287	10.6	11.2	11.9	11.9	
					Sepn tail	72.24	1.62	7.9	8.48	501	30.0	72.3	69.6	23.0	
					Feed (Cu-Pb ro conc)	100.00	3.90	7.98	8.80	1573	100.0	100.0	100.0	100.0	
C-6	SO_2 to RG AF 208 R343	1000 18 20 pH 5.1	No	Yes 80°C	Cu ro conc	36.16	4.16	3.33	3.52	1041	71.5	15.1	13.6	38.3	56.4
					Sepn tail	63.84	0.94	10.59	12.63	948.8	28.5	84.9	86.4	61.7	
					Feed (Cu-Pb ro conc)	100.00	2.10	7.97	9.34	982.1	100.00	100.0	100.0	100.0	
C-11	$K_2Cr_2O_7$, Dep "C" AF 208 AF 238 R343	400 125 18 18 30	Yes NaOH + heat 95°C	No	Cu ro conc	36.72	13.68	11.86	4.33	4407	68.8	17.8	26.3	55.5	51.0
					Sepn tail	63.28	3.60	31.6	7.0	2049	31.2	82.2	73.7	44.5	
					Feed (Cu-Pb cl conc)	100.00	7.29	24.35	6.02	2915	100.0	100.0	100.0	100.0	
C-12	$K_2Cr_2O_7$, Dep "C"	400 125 pH 8.0	Yes with acetone	No	Cu ro conc	38.53	17.24	25.50	4.63	5379.5	88.0	39.9	29.5	82.4	48.1
					Cu-Pb sepn tail	61.47	1.48	24.12	6.94	719.1	12.0	60.1	70.5	17.6	
					Feed (Cu-Pb cl conc)	100.00	7.55	24.65	6.05	2515	100.00	100.0	100.0	100.0	

On copper-lead cleaner concentrate the best copper-lead separation efficiency obtained was only 50.9% in test C-11. In this test sodium dichromate was used as the galena depressant in conjunction with Depramin "C" (carboxymethylcellulose). Also collectors were desorbed prior to separation step by conditioning with sodium hydroxide at a temperature of 95°C and discarding the solution. Note that this desorption step was very effective in removing collector coatings as the addition of a total of 60 g/t collector was subsequently required to float off a copper rougher concentrate. When the Test C-11 copper rougher concentrate was cleaned there was an increase in copper lead separation efficiency to 54.7% as indicated below in Table 11.

Table 11. Copper-lead separation results for Test C-11

Product	Wt %	Analysis				Distribution %				Sep eff %
		Cu %	Pb%	Zn %	Ag g/t	Cu	Pb	Zn	Ag	
Copper conc	25.58	17.60	6.70	3.70	5241.0	61.8	7.1	15.7	46.0	54.7
Copper cl tail	11.15	4.60	23.40	5.70	2478.0	7.0	10.7	10.6	9.5	
Sepn tail	63.27	3.60	31.60	7.00	2049.0	31.2	82.2	73.7	44.5	
Feed (Cu-Pb cl conc)	100.00	7.29	24.35	6.02	2915.0	100.0	100.0	100.0	100.0	
Copper ro conc	36.73	13.68	11.86	4.33	4407.0	68.8	17.8	26.3	55.5	51.0
Cu cl tail + sepn tail	74.42	3.75	30.37	6.81	2113.0	38.2	92.9	84.3	54.0	

To put the Test C-11 results in perspective they are compared with the Kidd Creek plant results in Table 12.

Table 12. Comparison of copper and lead concentrate grades and recoveries - Test c-11 versus Kidd Creek plant results*

Product	Analysis				Distribution %				Sep eff %
	Cu %	Pb%	Zn %	Ag g/t	Cu	Pb	Zn	Ag	
Copper conc - Test C-11	17.60	6.70	3.70	5247.0	50.1	5.7	0.6	33.8	44.4
Kidd Creek	18.42	9.62	4.30	5412.0	63.5	26.2	1.9	58.6	37.3
Lead conc - Test C-11**	3.75	30.37	6.81	2113.0	31.1	75.8	3.5	43.1	
Kidd Creek	3.84	15.79	7.46	1293.0	14.2	45.5	3.5	14.8	
Cu + Pb conc - Test C-11	7.30	24.35	6.02	2915.0	81.2	81.5	4.1	76.9	
Kidd Creek	11.77	12.79	5.93	3292.6	77.7	71.1	5.4	73.4	

* From Table 1

** Separation tail + copper cl tail

Note that the overall copper-lead separation efficiency obtained in Test C-11 is higher - 44.4% versus 37.3% for the Kidd Creek plant. Also higher combined copper, lead and silver recoveries in the copper and lead concentrates were obtained in Test C-11. A much higher proportion of the silver was recovered in the Kidd Creek copper concentrate - this is probably related to the different flotation techniques employed in Test C-11 and the Kidd Creek plant. The other large difference between the two sets of results is in the lead grade of the Kidd Creek plant lead concentrate which is only about half of that obtained in Test C-11.

It should be pointed out that this comparison of results is not strictly valid because: (1) the lead to copper ratio in the Test C-11 feed is much higher - 3.35:1 as 1.29:1 in the Kidd Creek plant feed and (2) batch results are being compared to plant results which include the effect of recycling cleaner tailings.

Flotation of a Secondary Zinc Concentrate from the Copper-Lead Separation Tailing

In Tests C-5 and 6 the copper-lead separation tailing was subjected to a zinc-lead separation step which consisted of floating off a secondary zinc concentrate at a pH of about 11 after activation with copper sulphate. Table 13 gives the results obtained for Test C-6.

Table 13. Results for flotation of a secondary zinc concentrate from the copper-lead separation tailing (Test C-6)

Product	Analysis				Distribution %			
	Cu %	Pb%	Zn %	Ag g/t	Cu	Pb	Zn	Ag
Sec zinc ro conc	0.49	12.79	36.40	1210.0	50.2	42.8	91.4	62.9
Zn-Pb sepn tail	0.24	8.45	1.70	353.0	49.8	57.2	8.6	37.1
Feed (Cu-Pb sepn tail)	0.32	9.88	13.17	636.3	100.0	100.0	100.0	100.0

As can be seen good selectivity was obtained against the pyrite to produce a relatively high grade secondary zinc rougher concentrate with high zinc recovery. However the galena was not effectively depressed by the high pH with the result that to a certain extent it was concentrated in the froth product. It is believed that had the secondary zinc rougher concentrate been subjected to multi-stage cleaning with lime, a high proportion of the galena would have been rejected to the cleaner tailings.

Except for copper-zinc separation of the bulk cleaner concentrate this technique was ineffective as a method for producing separate concentrates.

Deficiencies were as follows:

(1) Overall recoveries for bulk rougher flotation were appreciably lower than that obtained using the other flotation techniques. For example in test C-3 overall recoveries (in copper-lead rougher concentrate + zinc rougher concentrate) for Cu, Pb, Zn and Ag were 94.8, 92.3, 96.4 and 88.2% respectively compared to 88.8, 82.5, 93.7 and 74.2% for bulk rougher flotation in Test C-8. The zinc flotation technique employed for bulk rougher flotation, i.e. activation with copper sulphate in combination with-lime addition for a pH of 11+ generally gives very good selectivity against pyrite. This results in higher tailings losses because of the rejection of more of the pyrite-valuable sulphide middling particles to the tailing.

(2) The employment of a high pH in bulk rougher concentrate cleaning did not result in the rejection of most of the galena to the cleaner tailing as had been antisipated. Instead the lead content was splitbetween the concentrate and cleaner tailings in about the same proportion asthe weight of these products.

Cu-Pb-Zn Bulk Flotation Followed by Regrinding, Cleaning and Separation to Produce Separate Copper, Lead and Zinc Concentrates

The results obtained using this technique to produce separate concentrates is given in Table 14.

Table 14. Results for the production of separate concentrates from a bulk concentrate (Test C-8)

Product	Wt %	Analysis				Distribution %			
		Cu %	Pb%	Zn %	Ag g/t	Cu	Pb	Zn	Ag
Bulk Flotation									
Bulk ro conc	27.19	1.87	5.66	31.25	751.7	88.8	82.5	93.7	74.2
Bulk ro tail	72.81	0.09	0.45	0.78	97.6	11.2	17.5	6.3	25.8
Feed (calcd)	100.00	0.57	1.87	9.06	275.5	100.0	100.0	100.0	100.0
Bulk Rougher Concentrate Cleaning									
Bulk cl conc	63.88	2.48	5.94	44.25	873.7	84.4	67.1	90.5	74.3
Bulk cl tail (1+2)	36.12	0.81	5.15	8.25	536.1	15.6	32.9	9.5	25.7
Feed (calcd)	100.00	1.87	5.66	31.25	751.7	100.0	100.0	100.0	100.0
Copper-Zinc Separation on Bulk Cleaner Concentrate									
Copper conc	9.47	13.20	6.00	7.00	3562.2	50.3	9.5	1.5	38.5
Copper cl tail	12.82	7.70	9.20	18.70	1956.0	39.9	19.9	5.4	28.7
Sepn tail	77.71	0.31	5.40	53.00	368.3	9.8	70.6	93.1	32.8
Feed (calcd)	100.00	2.48	5.94	44.25	873.7	100.0	100.0	100.0	100.0
Copper ro conc	22.29	10.04	7.84	13.73	2638.4	90.2	29.4	6.9	67.2
Lead Flotation from Bulk Cleaner Tailings									
Lead ro conc 1	13.86	2.70	25.40	6.60	1781.0	46.1	68.3	11.1	46.0
Lead ro conc 2	11.78	1.50	6.30	11.10	917.6	21.8	14.4	15.9	20.2
Lead ro tail	74.36	0.35	1.20	8.10	243.5	32.1	17.3	73.0	33.8
Feed (calcd)	100.00	0.81	5.15	8.25	536.1	100.0	100.0	100.0	100.0
Lead ro conc 1+2	25.64	2.15	16.62	8.67	1384.3	67.9	82.7	27.0	66.[illegible]

(3) Because the bulk cleaner tailings contained only 27.1% of the lead in the feed the results obtained for lead flotation from this product cannot be d to be representative. Note that the addition of 100 g/t cyanide to the bulk ailing regrinding step prior to lead flotation was not sufficient to cause n of the copper which was concentrated along with lead in the froth product.

As note previously the only technique found to be effective in this test was nc separation of the bulk cleaner concentrate using sulphur dioxide in on with a high pulp temperature to depress sphalerite.

CONCLUSIONS AND RECOMMENDATIONS

The main problem encountered in treating this ore was the difficulty in achieving a satisfactory degree of selectivity between chalcopyrite and pyrite and galena and pyrite during rougher flotation. However, it was found that cyanide was an effective pyrite depressant when added to a regrinding step prior to cleaning of a copper-lead rougher concentrate. In rougher flotation the addition of 125 g/t NaCN to the primary grind appeared to have little depressing effect on pyrite. The most effective method for depressing pyrite was to first subject the ground ore to cyanidation for a 21-hour period at a NaCN strength of 1 g/L (equivalent to 2000 g NaCN/ton ore). Flotation from the cyanide residue produced a lead concentrate grading 47.4% lead with 62.8% recovery and a zinc concentrate grading 56.2% zinc with 85.3% recovery both after only one stage of cleaning. These results were achieved at a grind of 89.3% minus 44 µm. Only 5.1% of the zinc was rejected to the final tailing indicating that this grind is sufficient for optimum zinc recovery but for lead the tailing loss was 18.5% which could be due mostly to incomplete liberation from pyrite. In this test (C-9) most of the copper did not float with the zinc as was anticipated but was rejected to the final tailing. It is believed that it might have been possible to recover the copper by subjecting the lead rougher tailing to intense aeration and then floating off a copper concentrate prior to zinc flotation. Since an appreciable amount of silver is associated with the chalcopyrite the recovery of a copper concentrate would also result in an increase in overall silver recovery.

In view of the much improved selectivity against pyrite achieved by using the cyanidation scheme it is recommended that the investigation be continued along these lines. It may not be necessary to subject the ore to a long period of cyanidation. The addition of about 500 to 1000 g/t cyanide to primary grinding may give equivalent results. Also recommended is a quantitative mineralogical study of the ore and test products to determine the degree of association of pyrite with galena, chalcopyrite and silver minerals.

REFERENCES

1. Lakefield Research, Characterization and application of new organic depressants for the selective flotation of sulphide ores, Phase No. 1 Final Report, Table 7, page 14; DSS Contract No. 23440-6-9062/01-SQ.

APPENDIX

List of Abbreviations

SA	soda ash
CaO	lime
Cu	copper sulphate
Cr	sodium dichromate
CMC	carboxymethylcellulose
Acet	acetone

Cyanamid collectors:

317	sodium isobutyl xanthate
343	sodium isopropyl xanthate
31	Aerofloat 31
208	" 208
238	" 238
241	" 241
242	242
NaAF	Sodium aerofloat
3418	Aerophine 3418A

Frothers

250	Dowfroth 250
MIBC	Methyl isobutyl carbinol

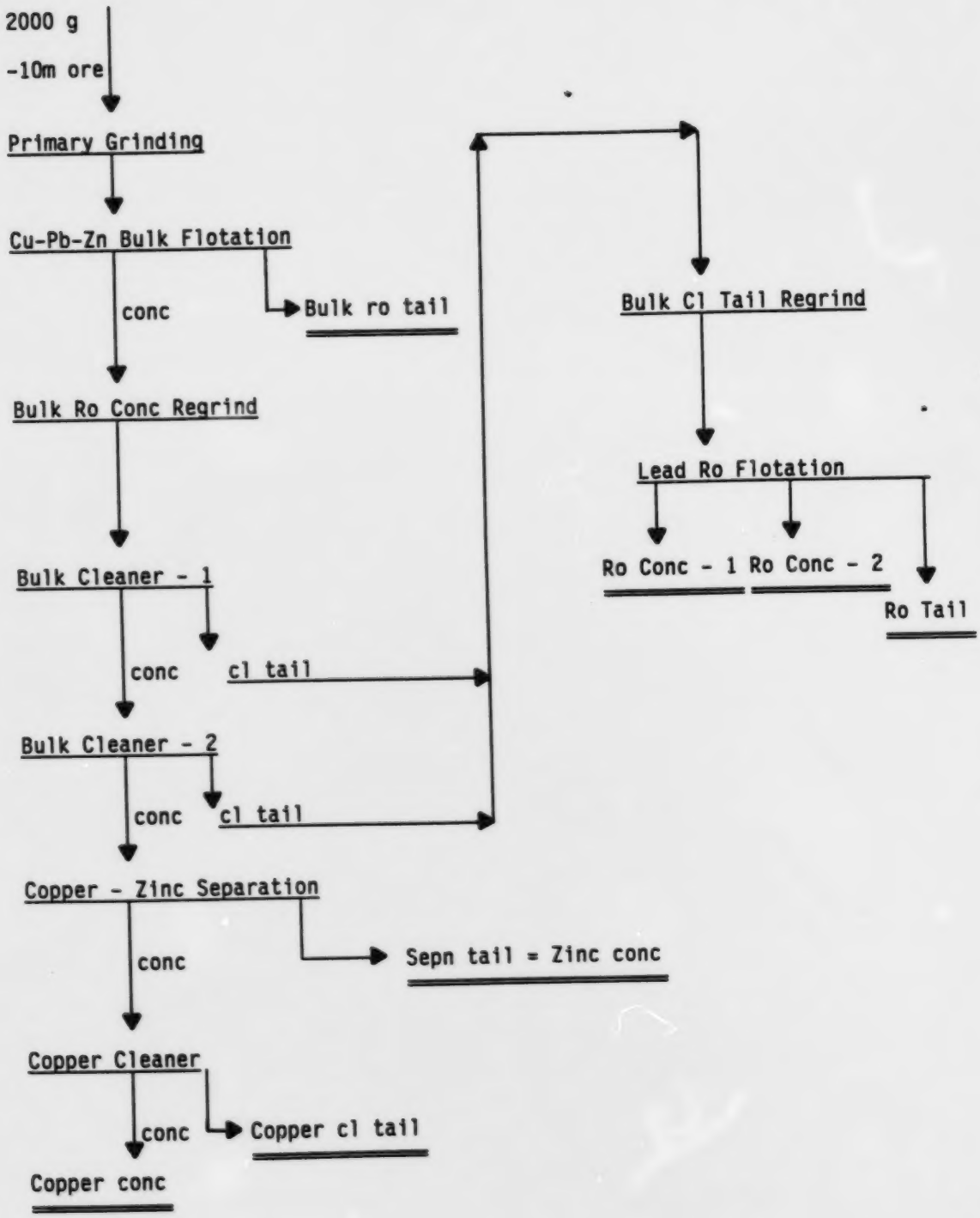

Fig. 1. Flowsheet for Test C-8

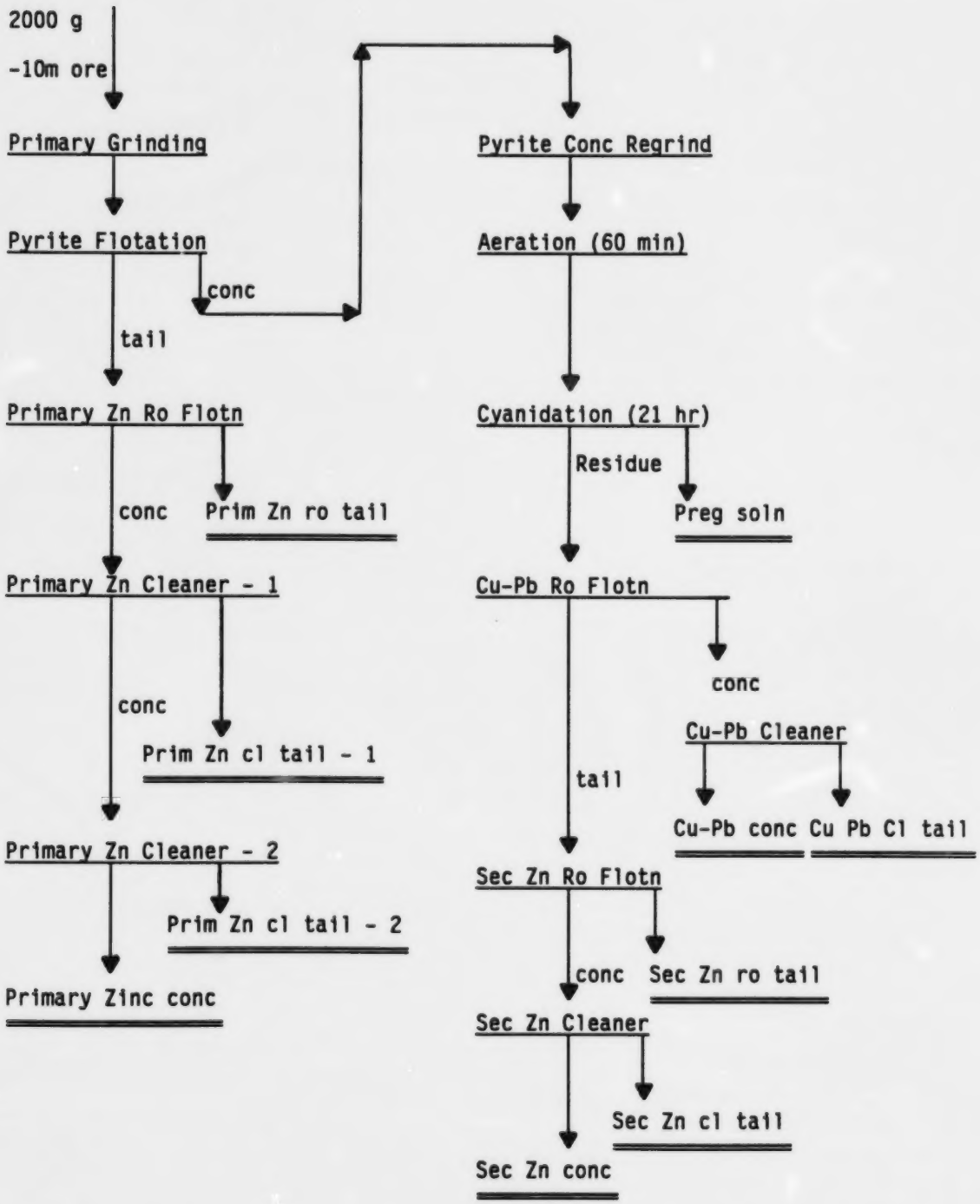

Fig. 2. Flowsheet for Test C-10

FLOTATION TEST REPORT

Test No: C-1	Sample: Kidd Creek "C" ore	Date: Feb. 25/88
Object of test: Sequential Cu-Pu-Zn selective flotation using SO_2		Charge: 2000 g
		Test by: GWL

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					SO_2	317	238	250	NaCN	SA	31	CaO	Cu	NaAF
Grinding	30	65	6.9	12-in. CM										
Filter - wash cake														
Grinding - 2	30	65	8.0	12-in. CM										
Aeration	20			Aerator										
Copper rougher														
Conditioning	5		5.3	1000-g cell	1875									
Float - 1	1					10	18	10						
Float - 2	1		5.4				18							
Filter Cu ro tail														
- wash cake														
Grinding - 3	30	65	7.8	12-in. CM		25			100	1000				
Lead rougher				1000-g cell										
Float - 1	1							10			20			
Float - 2	1		7.9			10								
Zinc rougher														
Conditioning	10		11.0	1000-g cell								900	1000	
Float - 1	1					5								52
Float - 2	1					5		10						13
Float - 3	2		10.6					10						13

REMARKS: Grinding - 1 with 19 kg 25 mm Steel balls
Grinding - 2 and 3 with 5.9 kg minus 16 mm ceramic balls

FLOTATION TEST REPORT

Test No: C-1	Sample: Kidd Creek "C" ore	Date: Feb. 25/88
Object of test:		Charge:
		Test by:

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					SO_2			250						
Copper cleaner	1		4.6	250-g cell	312									
Lead cleaner	1		7.9	250-g cell										
Zinc cleaner				500-g cell										
Float - 1	1		10.1											
Float - 2	1							10						

REMARKS:

METALLURGICAL BALANCE

Test No: C-1 | **Sample:** Kidd Creek "C" ore | **Date:** Feb. 25/88

PRODUCT	WT %	ANALYSIS				DISTRIBUTION %			
		Cu %	Pb %	Zn %	Ag g/t	Cu	Pb	Zn	Ag
Copper conc	7.68	5.94	14.80	6.00	2236.0	77.2	63.3	5.5	63.8
Copper cl tail	4.40	0.78	2.06	10.85	358.0	5.8	5.1	5.7	5.9
Lead conc	4.55	0.73	6.67	10.42	528.9	5.6	16.9	5.6	9.0
Lead cl tail	8.27	0.13	0.81	10.09	203.4	1.8	3.7	9.9	6.2
Zinc conc	11.49	0.17	0.39	50.48	141.0	3.3	2.5	68.9	6.0
Zinc cl tail	3.93	0.18	0.69	5.11	135.8	1.2	1.5	2.4	2.0
Zinc ro tail	59.68	0.05	0.21	0.28	32.0	5.1	7.0	2.0	7.1
Feed (calcd)	100.00	0.59	1.79	8.41	269.0	100.0	100.0	100.0	100.0
Calcd analyses									
Copper ro conc	12.08	4.06	10.16	7.77	1552.0	83.0	68.4	11.2	69.7
Lead ro conc	12.82	0.34	2.89	10.21	318.9	7.4	20.6	15.5	15.2
Zinc ro conc	15.42	0.17	0.46	38.92	139.6	4.5	4.0	71.3	8.0
Copper + lead ro conc	24.90	2.15	6.42	9.02	917.1	90.4	89.0	26.7	84.9
Lead ro tail	75.10	0.075	0.26	8.21	54.1	9.6	11.0	73.3	15.1

REMARKS:

FLOTATION TEST REPORT

Test No: C-2	Sample: Kidd Creek "C" ore	Date: Mar. 1/88
Object of test:	Sequential Cu-Pb-Zn selective flotation as in C-1 but employed single stage grinding	Charge: 2000 g
		Test by: GWL

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					SO_2	238	250	317	31	CaO	Cu	NaAF	NaCN	
Grinding	60	65	7.0	12-in. CM*										
Aeration	20		8.0	Aerator										
Copper rougher														
Conditioning	5		5.6	1000-g cell	2400									
Float - 1						18	10	10						
Float - 2								5						
Lead rougher float	1		7.7				10		20					
Lead scav float	1						10	10						
Zinc rougher														
Conditioning	10		10.9	1000-g cell						1500	1000			
Float - 1	1						10	10				52		
Float - 2	1						10	10				13		
Float - 3	2						10					13		
Copper cleaner - 1	1		5.0	250-g cell	312									
Copper cleaner - 2	1		4.8	250-g cell	100									
Lead cleaner														
Conditioning	5		8.9	250-g cell									100	
Float	1													

REMARKS: *With 19 kg 25 mm steel balls

METALLURGICAL BALANCE

Test No: C-2	Sample: Kidd Creek "C" ore	Date: Mar. 1/88

PRODUCT	WT %	ANALYSIS				DISTRIBUTION %			
		Cu %	Pb %	Zn %	Ag g/t	Cu	Pb	Zn	Ag
Copper conc	4.77	7.20	15.95	4.44	3680.5	57.9	40.5	2.4	58.4
Copper cl tail - 2	2.00	3.08	3.88	8.60	1470.1	10.4	4.1	1.9	9.8
Copper cl tail - 1	4.83	1.34	6.08	9.20	390.2	10.9	15.6	5.0	6.3
Lead conc	3.75	0.50	9.39	10.16	360.0	3.2	18.7	4.3	4.5
Lead cl tail	5.92	0.35	1.14	8.80	257.1	3.5	3.6	5.8	5.1
Lead scav conc	4.85	0.40	1.94	14.40	182.1	3.3	5.0	7.8	2.9
Zinc ro conc	17.44	0.21	0.60	36.26	112.1	6.1	5.6	70.8	6.5
Zinc ro tail	50.44	0.05	0.23	0.31	35.0	4.7	6.9	2.0	6.5
	100.00	0.59	1.88	8.93	300.7	100.0	100.0	100.0	100.0
Calculated analyses									
Copper ro conc	11.60	4.05	9.76	7.14	1929.4	79.2	60.2	9.3	74.5
Lead ro conc	9.67	0.41	4.34	9.33	297.1	6.7	22.3	10.1	9.6
Lead ro + scav conc	14.52	0.41	3.54	11.02	258.7	10.0	27.3	17.9	12.5
Copper ro + lead ro									
+ lead scav conc	26.12	2.02	6.30	9.30	1000.6	89.2	87.5	27.2	87.0
Lead ro tail	73.88	0.088	0.32	8.80	53.2	10.8	12.5	72.8	13.0

REMARKS:

FLOTATION TEST REPORT

Test No: C-3	Sample: Kidd Creek "C" ore	Date: Mar. 3/88
Object of test: Cu-Pb bulk flotation followed by Cu-Pb conc regrind and Cu-Pb separation		Charge: 2000 g
using dichromate + SO_2 + hot pulp		Test by: GWL

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					SA	NaCN	Zn	241	3418	250	Cr	So_2		
Grinding	60	65	9.0	12-in. CM*	1500	125	375	20						
Aeration	10		8.5	Aerator					10					
Copper-lead rougher				1000-g cell										
Float - 1	1									10				
Float - 2	1								5					
Float - 3	2									10				
Copper-lead rougher														
Conc regrind	30	50		8-in. CM**	250	50								
Copper-lead cleaner	2		9.2	500-g cell										
Copper-lead separation				250-g cell										
Conditioning	5 @ 70°C		8.0								200	75		
Copper ro float	1													
Copper cleaner			8.3								50			
Zinc rougher					CaO	Cu	NaAF	343	250					
Conditioning	10		11.0	1000-g cell	1250	1000								
Float - 1							52	10						
Float - 2							13							
Float - 3							13	10						

REMARKS: * Same charge as Test C-2

** with minus 16 mm ceramic ball charge occupying 1/2 volume of mill

METALLURGICAL BALANCE

Test No: C-3	Sample: Kidd Creek "C" ore	Date: Mar. 3/88

PRODUCT	WT %	ANALYSIS					DISTRIBUTION %				
		Cu %	Pb %	Zn %	Ag g/t		Cu	Pb	Zn	Ag	
Copper conc	2.95	14.36	22.17	5.44	4767.0		69.0	34.2	1.8	52.7	
Copper cl tail	1.37	3.00	27.12	8.40	1520.9		6.7	19.5	1.2	7.8	
Lead conc (sepn tail)	2.38	1.36	22.66	6.04	1115.6		5.3	28.2	1.6	9.9	
Copper-lead cl tail	11.10	0.42	0.95	10.40	247.5		7.6	5.5	12.7	10.3	
Zinc ro conc	18.34	0.21	0.51	39.30	109.7		6.2	4.9	79.1	7.5	
Zinc ro tail	63.86	0.05	0.23	0.52	49.4		5.2	7.7	3.6	11.8	
Feed (calcd)	100.00	0.61	1.91	9.11	267.1		100.0	100.0	100.0	100.0	
Calculated analyses											
Copper-lead ro conc	17.80	3.05	9.38	8.84	1210.4		88.6	87.4	17.3	80.7	
Copper-lead cl conc	6.70	7.41	23.36	6.26	2806.3		81.0	81.9	4.6	70.4	
Copper-lead ro tail	82.20	0.08	0.32	9.17	63.10		11.4	12.6	82.7	19.3	
Copper-lead separation											
Copper conc	44.01	14.36	22.17	5.44	4767.0		85.2	41.8	38.2	74.8	
Copper cl tail	20.51	3.00	27.12	8.40	1520.9		8.3	23.8	27.5	11.1	
Lead conc (sepn tail)	35.48	1.36	22.66	6.04	1115.6		6.5	34.4	34.3	14.1	
Feed (calcd)	100.00	7.42	23.36	6.26	2805.7		100.0	100.0	100.0	100.0	
Copper ro conc	64.52	10.75	23.74	6.38	3735.1		93.5	65.6	65.7	85.9	

REMARKS:

FLOTATION TEST REPORT

Test No: C-4	Sample: Kidd Creek "C" ore	Date: Mar. 8/88
Object of test: To try Dow collector F2B in sequential Cu-Pb-Zn selective flotation		Charge: 2000 g
		Test by: GWL

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					F2B	208	So_2	250	SA	NaCN	241	343	CaO	Cu
Grinding	90	65		12-in CM*	9									
Aeration	20			Aerator		18	1875							
Copper rougher														
Float - 1	1		6.3					10						
Float - 2	1					18								
Lead rougher														
Conditioning	20		7.4	1000 - g cell					1000	125				
Float - 1	1/2										15	5		
Float - 2	1/2		7.5								10			
Zinc rougher														
Conditioning	10		10.9	1000 - g cell							NaAF		2500	1000
Float - 1	1										52	15		
Float - 2	1											5		
Float - 3	2										26			
Zinc scavenger														
Float - 1	1/2											30		
Float - 2	1/2							10						

REMARKS: * Same charge as Test C-2

METALLURGICAL BALANCE

Test No: C-4	Sample: Kidd Creek "C" ore	Date: Mar. 8/88

PRODUCT	WT %	ANALYSIS					DISTRIBUTION %				
		Cu %	Pb %	Zn %	Ag g/t		Cu	Pb	Zn	Ag	
Copper ro conc - 1	4.33	0.80	2.78	9.02	2326.2		6.0	6.3	4.4	37.9	
Copper ro conc - 2	2.85	2.68	3.84	8.74	1435.9		13.2	5.7	2.8	15.2	
Lead ro conc	5.26	2.12	23.31	7.10	1111.5		19.3	63.5	4.2	21.6	
Zinc ro conc	16.06	1.85	1.12	38.10	179.7		51.4	9.3	68.3	10.7	
Zinc scav conc	4.12	0.28	1.08	34.95	194.7		2.0	2.3	16.0	3.0	
Zinc ro tail	67.38	0.07	0.37	0.58	46.6		8.1	12.9	4.3	11.6	
Feed (calcd)	100.00	0.58	1.93	8.96	270.0		100.0	100.0	100.0	100.0	
Calculated analyses											
Cu ro conc combined	7.18	1.55	3.20	8.91	1994.5		19.2	12.0	7.2	53.1	
Zinc not scav conc	20.18	1.53	1.11	37.46	182.7		53.4	11.6	84.3	13.7	

REMARKS:

FLOTATION TEST REPORT

Test No: C-5	Sample: Kidd Creek "C" ore	Date: Mar. 10/88
Object of test: To try Dow collector F2B in Cu-Pb bulk flotation followed by Cu-Pb and Zn-Pb separation on reground Cu-Pb ro conc		Charge: 2000 g
		Test by: GWL

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					F2B	343	CaO	3418	250	Cu	NaAF	Cr	208	
Grinding	90	65	9.5	12-in. CM*	9	5	250							
Aeration	10			Aerator				10						
Copper-lead rougher				1000-g cell										
Float - 1	1/2		8.0					5	10					
Float - 2	1/2								10					
Float - 3	1							5						
Float - 4	1							5						
Primary zinc rougher														
Conditioning	10		11.0	1000-g cell			1500			1000				
Float - 1						10					52			
Float - 2											13			
Float - 3											26			
Cu-Pb re conc regrind***	20	50		8-in. CM								400		
Copper-lead separation														
Cu ro conc 1- float 1	1/2		8.8	500-g cell									18	
Cu ro conc 1- float 2	1/2									10				
Cu ro conc 2- float	1/2													

REMARKS: * Same charge as in Test C-2
** Cu-Pb ro conc filtered, cake washed once with water and repulped with fresh water in regrind mill.

FLOTATION TEST REPORT

Test No: C-5	Sample: Kidd Creek "C" ore	Date: Mar. 10/88
Object of test:		Charge:
		Test by:

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					Cu	CaO	NaAF	343						
Zinc-lead separation														
Conditioning	10		10.8	500-g cell	500									
Sec zinc ro														
Float - 1	1/2						13	2.5						
Float - 2	1/2						13	2.5						

REMARKS:

METALLURGICAL BALANCE

Test No: C-5	Sample: Kidd Creek "C" ore	Date: Mar. 10/88

PRODUCT	WT %	ANALYSIS					DISTRIBUTION %				
		Cu %	Pb %	Zn %	Ag g/t		Cu	Pb	Zn	Ag	
Copper ro conc - 1	2.45	11.98	6.79	8.40	5287.4		47.4	10.4	2.3	48.1	
Copper ro conc - 2	1.06	4.91	10.61	12.44	2227.2		8.4	7.1	1.5	8.8	
Sec zinc ro conc	2.15	2.65	6.36	24.86	901.0		9.2	8.6	5.9	7.2	
Lead rougher conc *	6.99	1.30	8.49	3.46	378.5		14.7	37.2	2.7	9.8	
Prim zinc ro conc	15.81	0.34	1.31	40.82	116.6		8.7	13.0	71.4	6.8	
Zinc ro tail	71.54	0.10	0.53	2.05	72.7		11.6	23.7	16.2	19.3	
Feed (calcd)	100.00	0.62	1.60	9.03	269.4		100.0	100.0	100.0	100.0	
Copper-lead separation											
Copper ro conc - 1	19.36	11.98	6.79	8.40	5287.4		59.4	16.5	18.5	65.1	
Copper ro conc - 2	8.40	4.91	10.61	12.44	2227.2		10.6	11.2	11.9	11.9	
Cu-Pb sepn tailing	72.24	1.62	7.99	8.48	501.2		30.0	72.3	69.6	23.0	
Feed (Cu-Pb ro conc)	100.00	3.90	7.98	8.80	1572.8		100.0	100.0	100.0	100.0	
Zinc-lead separation											
Sec zinc ro conc	23.48	2.65	6.36	24.86	901.0		38.5	18.7	68.8	42.2	
Zn-Pb sepn tailing	76.52	1.30	8.49	3.46	378.5		61.5	81.3	31.2	57.8	
Feed (Cu-Pb sepn tail)	100.00	1.62	7.99	8.48	501.2		100.0	100.0	100.0	100.0	

REMARKS: *Zinc-lead separation tailing

FLOTATION TEST REPORT

Test No: C-6	**Sample:** Kidd Creek "C" ore	**Date:** Mar. 23/88
Object of test: Repeat of Test C-5 but employ So_2 + hot pulp for Cu-Pb separation instead of dichromate		**Charge:** 2000 g
		Test by: GWL

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					SO_2	343	250	208	Cr	Cu	CaO	NaAF		
Grinding) as														
Cu-Pb rougher) in														
Prim Zn rougher) C-5														
Cu-Pb ro conc regrind	20	50		8-in. CM	1000									
Copper-lead separation														
Conditioning (80°C)	15		5.2	500-g cell										
Cu ro conc 1 - float 1	1/2							18						
float 2	1/2													
Cu ro conc 2 - float	1					10	10							
Cu ro conc 3 - float	1					10	10							
Zinc-lead separation														
Conditioning - 1	5		11.1	500-g cell					400					
Conditioning - 2	10									500	950			
Sec Zn ro - float 1	1/2						30					26		
float 2	1/2											13		
float 3	1											26		

REMARKS: Removed solution and repulped with fresh water prior to Cu-Pb rougher conc regrind and zinc-lead separation.

METALLURGICAL BALANCE

Test No: C-6	Sample: Kidd Creek "C" ore	Date: Mar. 23/88

PRODUCT	WT %	ANALYSIS					DISTRIBUTION %				
		Cu %	Pb %	Zn %	Ag g/t		Cu	Pb	Zn	Ag	
Copper ro conc - 1	4.22	0.73	2.78	3.6	937.0		6.3	6.0	1.7	16.2	
Copper ro conc - 2	2.76	9.40	4.16	3.4	1200.0		53.0	5.9	1.1	13.6	
Copper ro conc - 3	1.23	6.50	16.99	7.8	3750.0		16.3	10.8	1.1	19.0	
Sec zinc ro conc	3.66	0.49	12.79	36.4	1210.0		3.7	24.1	15.2	18.2	
Zinc-lead sepn tail*	7.41	0.24	8.45	1.7	353.0		3.6	32.3	1.4	10.7	
Prim zinc ro conc	15.22	0.24	0.77	41.2	102.0		7.5	6.0	71.3	6.4	
Zinc ro tail	65.50	0.072	0.44	1.1	58.9		9.6	14.9	8.2	15.9	
Feed (calcd)	100.00	0.49	1.94	8.79	243.4		100.0	100.0	100.0	100.0	
Calcd analyses											
Comb copper ro conc	8.21	4.51	5.37	4.16	1446.9		75.6	22.7	3.9	48.8	
Comb zinc ro conc	18.88	0.29	3.10	40.27	316.8		11.2	30.1	86.5	24.6	
Copper-lead ro conc	19.28	2.10	7.97	9.34	982.1		82.9	79.1	20.5	77.7	
Copper-lead ro tail	80.72	0.10	0.50	8.66	67.1		17.1	20.9	79.5	22.3	

REMARKS: * Lead concentrate

METALLURGICAL BALANCE

Test No: C-6 | **Sample:** Kidd Creek "C" ore | **Date:** Mar. 23/88

PRODUCT	WT %	ANALYSIS					DISTRIBUTION %				
		Cu %	Pb %	Zn %	Ag g/t		Cu	Pb	Zn	Ag	
Copper-lead sepn											
Copper ro conc - 1	21.86	0.73	2.78	3.60	937.0		7.6	7.6	8.4	20.9	
Copper ro conc - 2	14.30	9.40	4.16	3.40	1200.0		63.9	7.5	5.2	17.5	
Copper ro conc - 3	6.40	6.50	16.99	7.80	3750.0		19.8	13.7	5.4	24.4	
Cu-Pb sepn tail	57.44	0.32	9.88	13.17	636.3		8.7	71.2	81.0	37.2	
Feed (Cu-Pb ro conc)	100.00	2.10	7.97	9.34	982.1		100.0	100.0	100.0	100.0	
Copper ro conc 1+2	36.16	4.16	3.33	3.52	1041.0		71.5	15.1	13.6	38.3	
Copper ro conc 3+sepn tail	63.84	0.94	10.59	12.63	948.8		28.5	84.9	86.4	61.7	
Feed (Cu-Pb ro conc)	100.00	2.10	7.97	9.34	982.1		100.0	100.0	100.0	100.0	
Zinc-lead sepn											
Sec Zn ro conc	33.08	0.49	12.79	36.4	1210.0		50.2	42.8	91.4	62.9	
Zn-Pb sepn tail	66.92	0.24	8.45	1.7	353.0		49.8	57.2	8.6	37.1	
Feed (Cu-Pb sepn tail)	100.00	0.32	9.88	13.17	636.3		100.0	100.0	100.0	100.0	

REMARKS:

FLOTATION TEST REPORT

Test No: C-8	Sample: Kidd Creek "C" ore	Date: April 20/88
Object of test: Cu-Pb-Zn bulk flotation followed by regrinding and separation to produce		Charge: 2000 g
Cu, Pb and Zn concentrates.		Test by: GWL

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					CaO	Cu	NaAF	343	242	250				
Grinding	50	65	8.3	12-in. CM	500									
Aeration - 1	10			Aerator	500	1000								
Aeration - 2	5		8.7	Aerator			52	15	20					
Bulk rougher				1000-g cell										
Float - 1	1/2		11.0		500			20						
Float - 2	1/2							10						
Float - 3	1							5						
Float - 4	1						26	5						
Float - 5	1									10				
Bulk ro conc regrind	30	50		8-in. CM	1000	250								
Bulk cleaner - 1				500-g cell										
Float - 1	1		11.4											
Float - 2	1							10						
Float - 3	1							10						
Float - 4	1							5						
Float - 5	2									10				
Bulk cleaner - 2				500-g cell										
Float - 1	2		11.2											
Float - 2	1									5				

REMARKS: Bulk rougher conc filtrate used for make-up water in regrinding.

FLOTATION TEST REPORT

Test No: C-8	Sample: Kidd Creek "C" ore	Date: April 20/88
Object of test:		Charge:
		Test by:

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					SO_2	208	SA	NaCN	Zn	242	343			
Cu-Zn separation of														
bulk cleaner conc														
Conditioning (86°C)	6		4.8	500-g cell	700									
Copper rougher														
Float - 1	1 1/2					9								
Float - 2	1/2					9								
Copper cleaner	1/2			250-g cell										
Lead flotation from														
comb bulk cl tail														
Regrind	10	50	10.8	8-in. CM			500	125						
Conditioning	5		9.7	500-g cell					500	10				
Ro conc 1 - float	1										5			
Ro conc 2 - float	1									10				

REMARKS: Fresh water used in regrinding of combined bulk cleaner tailings.

METALLURGICAL BALANCE

Test No: C-8	Sample: Kidd Creek "C" ore	Date: April 20/88

PRODUCT	WT %	ANALYSIS					DISTRIBUTION %				
		Cu %	Pb %	Zn %	Ag g/t		Cu	Pb	Zn	Ag	
Copper conc	1.64	13.20	6.00	7.00	3562.2		37.7	5.3	1.3	21.1	
Copper cl tail	2.23	7.70	9.20	18.70	1956.0		29.9	11.0	4.6	15.8	
Cu-Zn sepn tail*	13.50	0.31	5.40	53.00	368.3		7.3	39.0	78.9	18.1	
Lead ro conc - 1	1.36	2.70	25.40	6.60	1781.0		6.4	18.5	1.0	8.8	
Lead ro conc - 2	1.16	1.50	6.30	11.10	917.6		3.0	3.9	1.4	3.9	
Lead ro tail	7.30	0.35	1.20	8.10	243.5		4.5	4.7	6.5	6.5	
Bulk ro tail	72.81	0.09	0.45	0.78	97.6		11.2	7.6	6.3	25.8	
Feed (calcd)	100.00	0.57	1.87	9.06	275.5		100.0	100.0	100.0	100.0	
Bulk Cleaner											
Bulk cl conc	63.88	2.48	5.94	44.25	873.7		84.4	67.1	90.5	74.3	
Bulk cl tail (1+2)	36.12	0.81	5.15	8.25	536.1		15.6	32.9	9.5	25.7	
Feed (bulk ro conc)	100.00	1.87	5.66	31.25	751.7		100.0	100.0	100.0	100.0	

REMARKS: *Zinc concentrate

METALLURGICAL BALANCE

Test No: C-8	Sample: Kidd Creek "C" ore	Date: April 20/88

PRODUCT	WT %	ANALYSIS					DISTRIBUTION %				
		Cu %	Pb %	Zn %	Ag g/t		Cu	Pb	Zn	Ag	
Cu-Zn Separation											
on Bulk cl conc											
Copper conc	9.47	13.20	6.00	7.00	3562.2		50.3	9.5	1.5	38.5	
Copper cl tail	12.82	7.70	9.20	18.70	1956.0		39.9	19.9	5.4	28.7	
Sepn tail (zinc conc)	77.71	0.31	5.40	53.00	368.3		9.8	70.6	93.1	32.8	
Feed (bulk cl conc)	100.00	2.48	5.94	44.25	873.7		100.0	100.0	100.0	100.0	
Lead Flotation from											
Bulk Cleaner Tail											
Lead ro conc - 1	13.86	2.70	25.40	6.60	1781.0		46.1	68.3	11.1	46.0	
Lead ro conc - 2	11.78	1.50	6.30	11.10	917.6		21.8	14.4	15.9	20.2	
Lead ro tail	74.36	0.35	1.20	8.10	243.5		32.1	17.3	73.0	33.8	
Feed (bulk cl tail)	100.00	0.81	5.15	8.25	536.1		100.0	100.0	100.0	100.0	
Lead ro conc 1+2	25.64	2.15	16.62	8.67	1384.3		67.9	82.7	27.0	66.2	

REMARKS:

FLOTATION TEST REPORT

Test No: C-9	Sample: Kidd Creek "C" ore	Date: May 2/88
Object of test: Cyanidation of ore followed by lead and zinc flotation of cyanide residue		Charge: 2000 g
		Test by:

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					CaO	NaCN	343	241	MIBC	Cu	NaAF			
Grinding	50	65		12-in CM										
Aeration	60			Aerator	2000									
Cyanidation*	21 hr	33		1000-g cell	400	3300								
Lead rougher														
Conditioning	10		8.5	1000-g cell			10	10						
Float - 1	1/2								10					
Float - 2	1 1/2							10						
Lead cleaner	1			250-g cell										
Zinc rougher				1000-g cell										
Conditioning	10		10.9		1050					1500				
Float - 1	1/2						15				52			
Float - 2	1/2						5							
Float - 3	1										26			
Float - 4	2								5					
Zinc cleaner				500-g cell										
Float - 1	1		10.0											
Float - 2	1								5					

REMARKS: * NaCN strength start 1.0 g/L, pH 11.0

" " end 0.2 g/L, pH 8.1

METALLURGICAL BALANCE

Test No: C-9 | **Sample:** Kidd Creek "C" ore | **Date:** May 2/88

PRODUCT	WT %	ANALYSIS					DISTRIBUTION %				
		Cu %	Pb %	Zn %	Ag g/t		Cu	Pb	Zn	Ag	
Pregnant soln.	-	-	-	-	149.9*		-	-	-	56.1	
Lead conc.	2.66	0.48	47.40	6.50	1716.0		2.2	62.8	1.9	17.1	
Lead cl tail	2.84	1.20	7.10	8.00	523.6		6.0	10.0	2.4	5.6	
Zinc conc	14.11	0.34	0.86	56.20	130.6		8.4	6.0	85.3	6.9	
Zinc cl tail	4.51	1.50	1.20	10.90	135.9		11.8	2.7	5.3	2.3	
Zinc ro tail	75.88	0.54	0.49	0.62	42.3		71.6	18.5	5.1	12.0	
Feed (calcd)	100.00	0.57	2.01	9.29	267.1		100.0	100.0	100.0	100.0	
Calcd analyses											
Lead ro conc	5.50	0.85	26.59	7.27	1100.3		8.2	72.8	4.3	22.7	
Zinc ro conc	18.61	0.62	0.94	45.25	132.0		20.2	8.7	90.6	9.2	

REMARKS: * Expressed as g/t feed

FLOTATION TEST REPORT

Test No: C-10	Sample: Kidd Creek "C" ore	Date: May 16/88
Object of test: Flotation and cyanidation of pyrite conc. followed by copper-lead and zinc flotation from cyanide residue.		Charge: 2000 g
		Test by: G.W.L.

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					208	242	343	250	CaO	Cu	NaAF			
Grinding	50	65		12-in CM	18	20	20							
Pyrite flotn				1000-g cell										
Conditioning	5													
Float - 1	1						15	10						
Float - 2	1						15							
Float - 3	1						15							
Float - 4	1						15							
Float - 5	1						15							
Primary zinc ro				1000-g cell										
Conditioning	10		11.1						1050	1000				
Float - 1	1							10						
Float - 2	1										26			
Float - 3	2							10			26			
Primary zinc cl - 1			10.0	500-g cell										
Float - 1	1													
Float - 2	1							10						
Primary Zinc cl - 2														
Float - 1	1/2		9.4	250-g cell										
Float - 2	1							5						

REMARKS:

FLOTATION TEST REPORT

Test No: C-10	Sample: Kidd Creek "C" ore	Date: May 16/88
Object of test:		Charge:
		Test by:

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					CaO	NaCN	241	343	250	Cu	NaAf			
Ryrite conc.														
Regrinding	30	50		8-in CM										
Aeration	60			Aerator	2000									
Cyanidation*	24 hr	33		1000-g cell	3000	2000								
Cu - Pb rougher flotn														
Aeration	30		8.2	Aerator			10	10						
Conditioning	2			1000-g cell			10	20						
Float - 1	1													
Float - 2	1							20						
Float - 3	1							20	10					
Sec Zn ro flotn														
Conditioning	10		11.0		350					500				
Float - 1	1													
Float - 2	2										26			
Cu-Pb cleaner	2		8.2	250-g cell										
Sec Zn cleaner	2			250-g cell										

REMARKS: * NaCN strength start 1.0 g/L, pH 11.84 Pyrite conc filtered and cake washed before and NaCN strength end 0.2 g/L, pH 8.5 after regrinding and dried prior to cyanidation.

METALLURGICAL BALANCE

Test No: C-10	Sample: Kidd Creek "C" ore	Date: May 16/88

PRODUCT	WT %	ANALYSIS					DISTRIBUTION %				
		Cu %	Pb %	Zn %	Ag g/t	Fe tot	Cu	Pb	Zn	Ag	Fe tot
Pregnant soln*	-	0.012	-	0.006	87.8	-	2.0	-	0.1	35.5	-
Copper-lead conc	7.91	3.90	13.30	8.30	946.7	25.20	51.8	55.8	7.1	30.3	10.1
Copper-lead cl tail	8.72	0.79	3.20	9.30	182.0	33.40	11.6	14.8	8.8	6.4	14.7
Sec zinc conc	5.64	0.93	2.80	46.00	387.9	12.80	8.8	8.4	28.2	8.9	3.6
Sec zinc cl tail	2.19	1.70	2.60	3.50	174.5	30.90	6.2	3.0	0.8	1.6	3.4
Sec zinc ro tail	17.58	0.35	1.10	0.66	74.7	37.20	10.3	10.3	1.3	5.3	33.0
Prim zinc conc	8.15	0.22	0.33	57.30	101.5	7.10	3.0	1.4	50.7	3.3	2.9
Prim zinc cl tail - 1	0.50	0.64	1.40	11.10)	117.8	17.90	0.5	0.4	0.6)	1.6	0.5
Prim zinc cl tail - 2	2.84	0.29	0.69	3.00)		16.00	1.4	1.0	0.9)		2.3
Prim zinc ro tail	46.47	0.056	0.20	0.29	37.9	12.60	4.4	4.9	1.5	7.1	29.5
Feed (calcd)	100.00	0.60	1.89	9.21	247.2	19.82	100.0	100.0	100.0	100.0	100.0
Calcd analyses											
Pyrite conc	42.04	1.29	4.14	10.13	517.0	30.55	90.7	92.3	46.3	88.0	64.8
Prim + sec zinc conc	13.79	0.51	1.34	52.68	218.6	9.43	11.8	9.8	78.9	12.2	6.5

REMARKS: * Analyses expressed in terms of equivalent content in feed.

FLOTATION TEST REPORT

Test No: C-11	Sample: Kidd Creek "C" ore	Date: June 7/88
Object of test: Cu-Pb bulk flotation followed by Cu-Pb separation employing (1) collector		Charge: 2 X 2000 g
desorption using NaOH + heat and (2) dichromate + CMC		Test by: GWL

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					SA	NaCN	Zn	241	3418	250	NaOH	Cr	CMC	
Grinding	50	65		12-in CM	1500	125	375	20						
Aeration	10			Aerator					10					
Copper-lead rougher				1000-g cell										
Float - 1	1		8.3							10				
Float - 2	1								5					
Float - 3	2									10				
Cu-Pb ro conc regrind	30	50		8-in CM	250	50								
Cu-Pb cleaner - 1				500-g cell										
Float - 1	1/2		9.5											
Float - 2	1 ½									10				
Cu-Pb cleaner - 2				250-g cell										
Float - 1	1													
Float - 2	1									5				
Cu-Pb separation														
Desorption at 95°C	10		10.0	500-g cell							2000			
Conditioning	5		8.1									400	125	
Copper rougher *	3									10				
Copper cleaner	2			250-g cell										

REMARKS: (1) Cu-Pb ro conc filtered and repulped with fresh water in regrind mill
(2) Cu-Pb cl conc filtered prior to and after desorption step - fresh water to Cu-Pb separation
* Collectors added: 208-18, 343-30, 238-18 g/t

METALLURGICAL BALANCE

Test No: C-11	Sample: Kidd Creek "C" ore	Date: June 7/88

PRODUCT	WT %	ANALYSIS					DISTRIBUTION %				
		Cu %	Pb %	Zn %	Ag g/t		Cu	Pb	Zn	Ag	
Copper conc	1.57	17.60	6.70	3.70	5247.0		50.1	5.7	0.6	33.8	
Copper cl tail	0.69	4.60	23.40	5.70	2478.0		5.7	8.8	0.4	7.0	
Cu-Pb sepn tail	3.89	3.60	31.60	7.00	2049.0		25.4	67.0	3.0	32.7	
Cu-Pb cl tail - 2	0.69	0.78	1.60	12.20	275.4		1.0	0.6	1.0	0.8	
Cu-Pb cl tail - 1	8.71	0.59	1.10	10.80	195.3		9.3	5.2	10.5	7.0	
Cu-Pb rougher tail	84.45	0.056	0.28	9.00	54.5		8.5	12.7	84.5	18.7	
Feed (calcd)	100.00	0.55	1.84	9.00	244.1		100.0	100.0	100.0	100.0	
Calculated analyses											
Cu-Pb ro conc	15.55	3.25	10.31	8.97	1273.9		91.5	87.3	15.5	81.3	
Cu-Pb cl conc	6.15	7.29	24.32	6.01	2915.0		81.2	81.5	4.0	73.5	
Copper-lead separation											
Copper conc	25.58	17.60	6.70	3.70	5247.0		61.7	7.1	15.7	46.0	
Copper cl tail	11.15	4.60	23.40	5.70	2478.0		7.0	10.7	10.6	9.5	
Lead conc (sepn tail)	63.27	3.60	31.60	7.00	2049.0		31.3	82.2	73.7	44.5	
Feed (Cu-Pb cl conc)	100.00	7.29	24.32	6.01	2915.0		100.0	100.0	100.0	100.0	

REMARKS:

FLOTATION TEST REPORT

Test No: C-12	**Sample:** Kidd Creek "C" ore	**Date:** June 22/88
Object of test: Repeat of C-11 but use acetone to desorb collector instead of NaOH		**Charge:** 2 X 2000 g
		Test by: GWL

OPERATION	Time min	% S	pH	Unit used	REAGENTS ADDED, g/t									
					Acet	Cr	CMC	250	Cu	CaO	343	NaAF		
Grinding)														
Cu-Pb rougher) as														
Regrinding) in C-11														
Cu-Pb cleaners)														
Cu-Pb separation														
Desorption	10			500-g cell	100 cc									
Conditioning	10		8.1	500-g cell		400	125							
Copper rougher	3							10						
Conditioning	10		11.1						500	550				
Copper scavenger	1/2							10			40	26		

REMARKS: Pulp filtered and filter cake washed free of solution before and after desorption steps.

METALLURGICAL BALANCE

Test No: C-12	Sample: Kidd Creek "C" ore	Date: June 22/88

PRODUCT	WT %	ANALYSIS				DISTRIBUTION %			
		Cu %	Pb %	Zn %	Ag g/t	Cu	Pb	Zn	Ag
Copper conc	1.93	20.80	22.90	3.76	6177.0	68.9	23.1	0.8	46.3
Copper cl tail	0.55	4.76	34.60	7.67	2581.0	4.5	9.9	0.5	5.5
Copper scav conc	1.03	1.16	29.00	10.77	1814.0	2.0	15.6	1.3	7.3
Cu-Pb sepn tail	2.93	1.60	22.40	5.60	965.5	8.1	34.3	1.8	11.0
Cu-Pb cl tail - 2	1.09	0.71	1.50	11.86	414.4	1.3	0.9	1.4	1.7
Cu-Pb cl tail - 1	10.31	0.46	0.95	10.67	262.0	8.1	5.1	12.4	10.5
Cu-Pb ro tail	42.16	0.05	0.26	8.87	55.7	7.1	11.1	81.8	17.7
Feed (calcd)	100.00	0.58	1.92	8.91	257.7	100.0	100.0	100.0	100.0
Calculated analyses									
Cu-Pb ro conc	17.84	3.04	9.54	9.08	1187.9	92.9	88.9	18.2	82.3
Cu-Pb cl conc	6.43	7.58	24.67	6.06	2799.7	83.5	82.9	4.4	70.1
Copper-lead separation									
Copper conc	29.93	20.8	22.90	3.76	6177.0	82.5	27.8	18.6	66.0
Copper cl tail	8.60	4.76	34.60	7.67	2581.0	5.4	12.1	10.9	7.9
Copper scav conc	15.97	1.16	29.00	10.77	1814.0	2.4	18.8	28.4	10.4
Lead conc (sepn tail)	45.50	1.60	22.40	5.60	965.5	9.7	41.3	42.1	15.7
Feed (Cu-Pb cl conc)	100.00	7.58	24.67	6.06	2799.7	100.0	100.0	100.0	100.0

REMARKS: